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DEPARTMENT OF COMMERCE

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MINING PRACTICE AT THE CHINO MINES, NEVADA CONSOLIDATED COPPER CO., SANTA RITA, N. MEX.

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MINING PRACTICE AT THE CHINO MINES, NEVADA CONSOLIDATED COPPER CO., SANTA RITA, N. MEX.[1]

By H. A. Thorne[2]

INTRODUCTION

This paper describing the mining practices at the Chino mines at Santa Rita, Grant County, N. Mex., is one of a series being prepared by the Bureau of Mines on mining practices, methods, and costs in the various mining districts of the United States.

The Nevada Consolidated Copper Co., Chino mines, operates an open-cut copper mine at Santa Rita, Central mining district, Grant County, N. Mex. This mine is located about 15 miles east and slightly north of Silver City, the county seat. The ore is transported in standard railroad cars by the Atchison, Topeka & Santa Fe Railway Co., approximately 10 miles to Hurley where it is milled. The concentrates are shipped approximately 170 miles over the same railroad to the smelter of the American Smelting and Refining Co., at El Paso, Tex. (fig.1)

ACKNOWLEDGMENTS

The author acknowledges the assistance of Frank E. Thurber, chief engineer, Gerald J. Ballmer, geologist, and others of the staff of Chino mines in the gathering of material necessary to the preparation of this paper.

HISTORY OF DISTRICT

Whether the native copper of this basin was known to any prehistoric race and utilized by them is not certain, but it is stated that the cropping of native copper near the site of the old Romero shaft was shown to Colonel Jose Manuel Carasco, an officer in charge of one of the Spanish military posts in New Mexico in 1800, by an Indian chief with whom he had friendly relations.

Colonel Carasco interested Don Manuel Francisco Elguea, of Chihuahua, a wealthy merchant, banker, and subdelegate to the Spanish court, and a concession to the area was obtained under the Spanish law.

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1 - The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6412."

2 - One of the consulting engineers, U. S. Bureau of Mines, and superintendent of mines, Chino mines, Nevada Consolidated Copper Co.

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In 1804 Elguea purchased Carasco's interest in the property and immediately started to work. Elguea, through his influence, made a contract with the government to supply the Mexican province with the copper required in its coinage, the government agreeing to take the whole amount of copper produced.

The region being infested, as it was until the nineties, by the Apache Indians who were not subjugated until almost exterminated, it was necessary to erect a fort and maintain a military force to protect the camp and provide convoys. The fort was triangular in shape, consisting of a heavily built adobe tower at each of the apexes, joined by heavy walls. One of these towers is standing to-day.

Rough smelting works were erected and operations proceeded in spite of constant annoyances from the Apaches. The copper was transported under convoy by mule trains to Chihuahua and the City of Mexico.

In 1809 Elguea died and Juan Onis operated the property as superintendent for the heirs of Elguea.

The first American to visit the district was James Pattie, a trapper, hunter, and explorer. Pattie and his son leased the mine for five years, agreeing to pay $1,000 per year, and worked the property until 1827. Following the abandonment of the property by Pattie the mine was operated by a French resident of Chihuahua.

About 1834 a certain Robert McKnight took possession of the property and continued operations until 1836. Kit Carson was in the employ of McKnight as a teamster. In 1840 Leonardo Sesquieros took over the property and despite many obstacles successfully operated it until the late fifties, when the camp was abandoned owing to the depredations of the Apaches.

For several months in 1851 the American commissioners of the Boundary Survey made their headquarters at Santa Rita. Sometime before 1860 two men, named Sweet and Lacorte, took possession of the property, probably locating claims under the United States mining laws and were later joined by two others.

In 1862 General Sibley, in command of the Confederate forces from Texas, held this region for a short time; evidence of their occupancy was found in 1910 when a part of the old fort was razed to make room for the present machine shop.

In 1873, when the Apaches had been partly subdued, M. D. Hayes, connected with the first smelting works in Colorado, acting in behalf of Denver people, succeeded in buying the mining claims from the previous owners. He attempted to perfect the title by United States patents, in spite of the fact that patents had been refused by the Commissioner of the Land Office on April 21, 1870. The application of Hayes was denied on April 15, 1873, by Commissioner Drummond, who reiterated the decision of his predecessor. Appeals were made to the Secretary of the Interior, who in November 6, 1873, reaffirmed the previous decisions, ruling that the claimants had in no sense any right to the ground, which had

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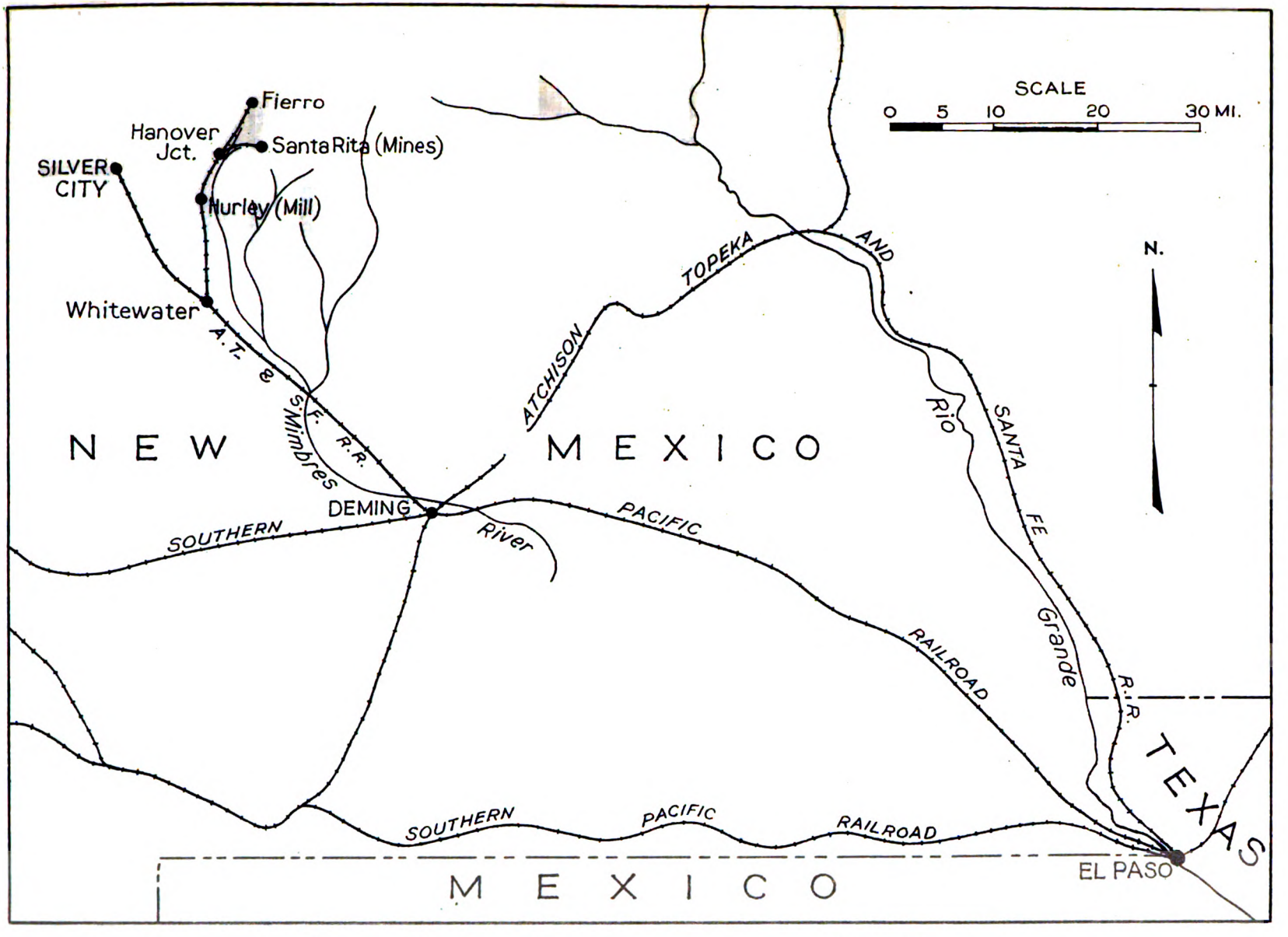

Figure 1.- Railroad map, southern New Mexico

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been known to the department for more than half a century as belonging to Elguea. In the meantime Hayes and his associates had been tracing the heirs of the Elguea estate, who were found to be scattered through Mexico and Europe. In October, 1873, title was obtained from these heirs.

Steps were then taken to complete the title and avoid the possibility of further question by locating the ground under the United States mining laws. Finally, in addition to the title obtained from the Elguea heirs, United States patents were obtained.

Hayes operated the property for a number of years, erecting a small smelting plant and shipping some high-grade ore and black copper to the nearest railroad station in Colorado from whence it was forwarded to the eastern coast.

In 1880 the property was sold to J. Parker Whitney of Boston. Shortly after, in 1881, a concentrating mill was built near the Romero shaft sunk by Hayes. Stamps were used for crushing the ore. Foundations and a few scattering pieces of the old machinery remained on the ground up to 1910. The property was divided under Whitney's ownership. The Bonanza Development Co. controlled the property on the south side of Santa Rita Creek, and the Santa Rita Copper and Iron Co. controlled the property on the north side of the creek.

In 1897, Whitney, who controlled both of these companies, gave a lease and bond to the Hearst estate, then operating a mine at Pinos Altos and a smelter in Silver City. In 1899 Whitney sold the property to a group of capitalists who were intimately associated with the Amalgamated Copper Co. A new corporation was organized, known as the Santa Rita Mining Co.

In 1891 the railroad connection from Whitewater, on the Rincon, Deming, and Silver City branch of the Santa Fe, was completed to Hanover and Fierro, while the branch from San Jose, or Hanover Junction, to Santa Rita was completed early in 1899.

Under the new owners development work was instituted and a lessee's mill was taken over by the company. At the same time lessee's operations were encouraged and a considerable part of the revenue came from royalties paid.

Near the end of the year 1904 the Hermosa Copper Co., a subsidiary of the General Electric Co., became interested in the property lying outside the Santa Rita basin, and active development work was undertaken. In the latter part of 1905 the Hermosa Copper Co. made arrangements whereby an examination was made of the holdings of the Santa Rita Mining Co. The work of the examination required about nine months. It was not until 1908 that an active interest was taken in this ore deposit.

The Chino Copper Co., organized in June, 1909, purchased the property. The principal interests connected with the Chino Copper Co. were also associated with the Utah Copper Co., and a vigorous development campaign was instituted. By the fall of 1910 steam shovels were at work, and a year later the first ore was milled at Hurley. In 1924 the Chino Copper Co. was absorbed by the Ray Consolidated Copper Co., and in 1926 the entire properties of the Ray Consolidated Copper Co. were purchased by the Nevada Consolidated Copper Co. operating at McGill, Nev.

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B. S. Butler in the Mineral Resources of the United States, 1912, is quoted as follows:

> "From 1845 to 1912 New Mexico has a recorded output of 124,353,963 pounds of copper, or 0.71 per cent of the output of the country since 1845. The principal production has been from the districts in Grant County."

"The districts in Grant County" may practically be interpreted as the Santa Rita district.

No recorded figures are available for the years previous to 1845, but from the various works in which the Santa Rita del Cobre grant is mentioned, it is estimated that previous to 1845 there were produced not less than 41,000,000 pounds of copper, which would make the production to 1912, 165,353,963 pounds. Accurate data is available since 1912, so that, except for a small percentage of error, the production of the Santa Rita mines can be placed at 1,204,000,000 pounds of copper up to the year 1930.

GEOLOGY

In Professional Paper 68 of the United States Geological Survey, entitled "Ore Deposits of New Mexico," published in 1910, is found a description on page 305, et. seq., of the Santa Rita and Hanover district in the Central mining district, Grant County, N. Mex. This article is written by L. C. Graton.

In 1916 the United States Geological Survey published the Silver City Folio, New Mexico, by Sidney W. Paige. The Silver City quadrangle includes the Central mining district, of which the Santa Rita area is a part. The Santa Rita mining district is located just north of the center of the eastern portion of the quadrangle. Beginning on page 16 of this folio is a description of the Santa Rita ore bodies indicating the progress made in detail studies following the description written in 1906. The United States Geological Survey has planned a detailed sheet known as the "Santa Rita Special" covering the results of further studies of the area. It is expected that this monograph, or professional paper will be published in the course of the next few months. Pending the publication of this final detailed study by the United States Geological Survey, the author approaches a description of the geology immediately connected with the ore deposits at Santa Rita with caution, although he has a general visualization of the historical, structural, and economic geology of the area.

As has been stated in the foregoing references, the structure in the Santa Rita area is most complex, especially as to details. The sediments of upper Paleozoic and Mesozoic ages were intruded during the Cretaceous period by igneous magmas in both concordant and crosscutting masses. This rock is generally accepted as a quartz diorite porphyry and is designated on the United States Geological Survey maps as "Kdp". The concordant masses of quartz diorite porphyry have assumed large proportions.

Following the appearance of the quartz diorite porphyry, doming and further fracturing of the sediments with their intercalated sill was caused by the intrusion of a stock of granodiorite porphyry. This stock formed a large

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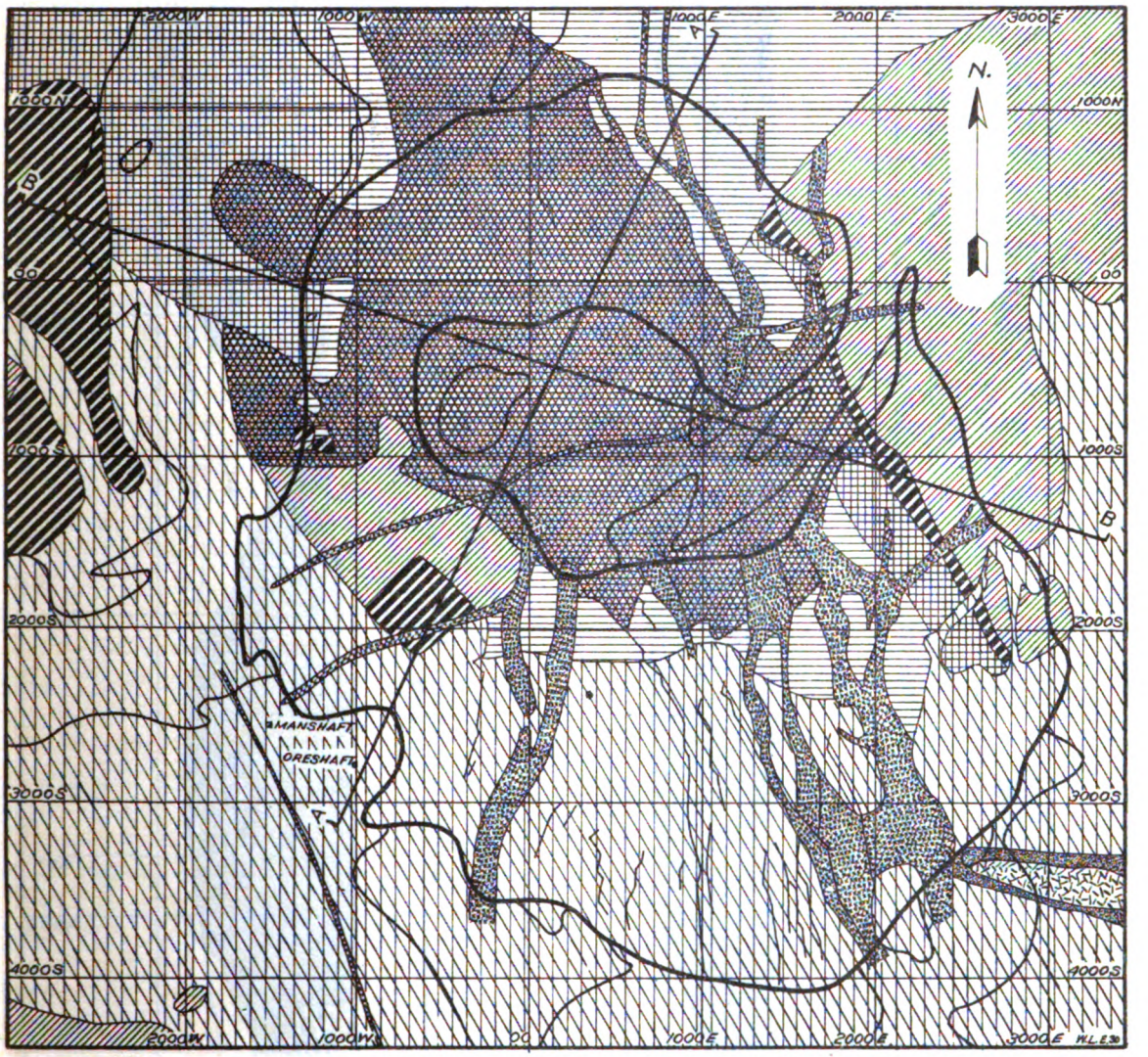

LEGEND

Diorite Porphyry	Dikes	Fierro Limestone	Gravel, sand & tuff	Abo? Red Beds
Granodiorite Porphyry	Colorado Shales	Beartooth Quartzite	Rhyolite	Pit Outline

SCALE

0 500 1000 1500 2000

Figure 2.- Surface geology

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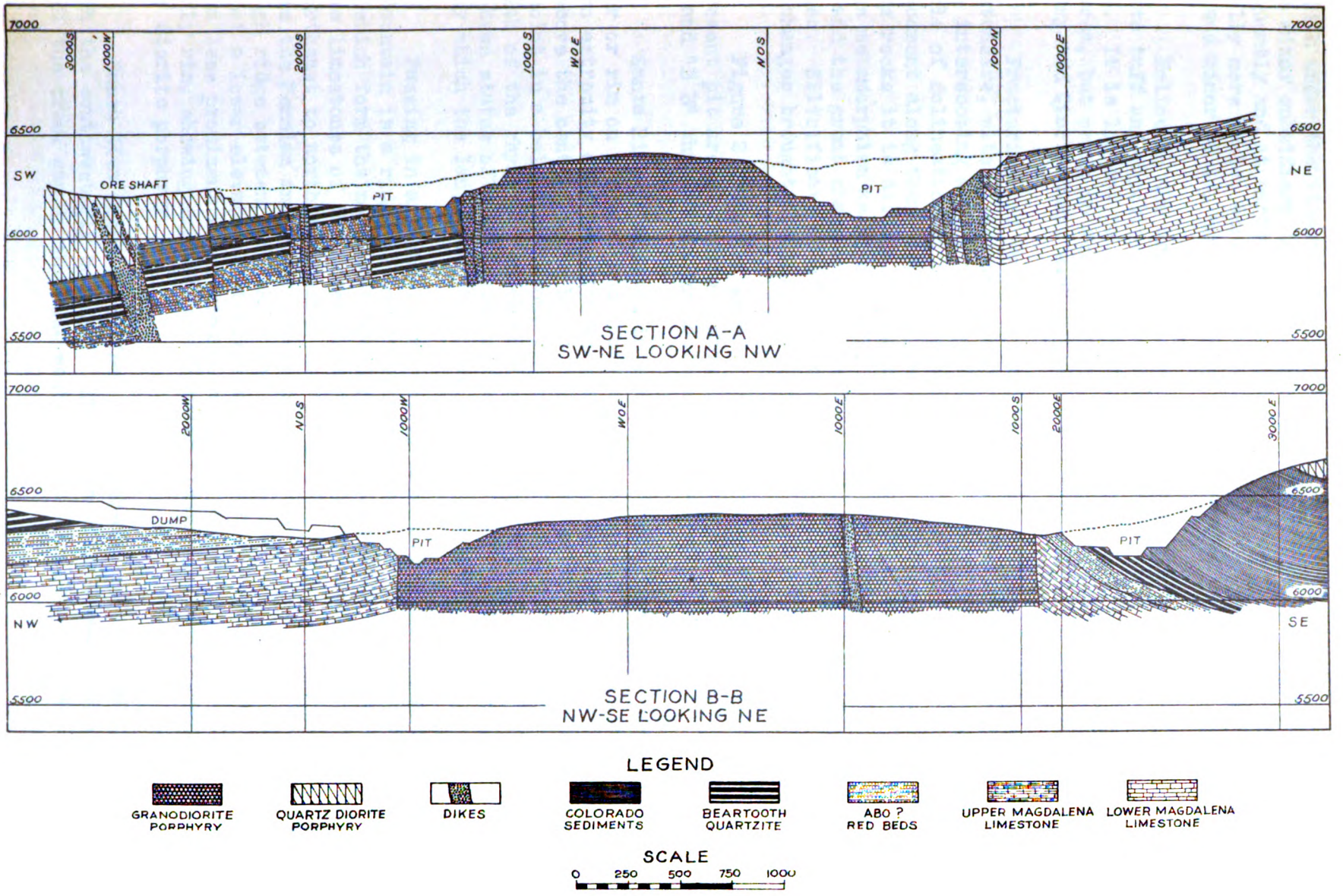

Figure 3.- Geologic cross sections

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batholith under the limestone series outcropping on either side of Hanover creek with a minor subsidiary dome now exposed in the center of the Santa Rita basin. Subsequently and at various times magmas of slightly different composition, but generally more basic, were forced into the resulting rock complex in the form of dikes and minor sills, causing further faulting and fracturing.

Following the Cretaceous and during Tertiary times immense flows of rhyolite tuff and rhyolite covered the northern portion of the Silver City quadrangle. It is logical to think that these flows at one time covered the Santa Rita area, but owing to further orogenic movements were eroded in the late Tertiary and Quarternary, exposing the underlying older rocks.

Fracturing and faulting in both northwest and northeast directions are extensive, with minor, more nearly north and south displacements. The major series intersecting around the granodiorite core make a complex structure impossible of delineation on small-scale drawings. Few faults appear to have much displacement along them, although because of alterations of both sediments and igneous rocks it is difficult to make a determination of such displacements. Igneous metamorphism as applying to the Santa Rita area has been of minor importance and the great changes have been brought about by meteoric and hydrothermal agencies. Silicification, sericitization, and kaolinization account for the major changes brought about in the rocks.

Figure 2 gives the general geology of the surface and an outline of the present pit area. Figure 3 shows geologic cross sections on two lines "A A" and "B B" through the granodiorite central core.

Santa Rita is located in a distinct basin, having a high abrupt barrier or rim on the east and southeast formed by the rhyolite of the northwestern extremity of Santa Rita Mountain which rises to a height of some 1,600 feet above the center of the basin. The escarpment of Santa Rita Mountain comes to a point in a bold bluff about 75 feet high. At the base of this bluff is a monolith of the rhyolite, which when viewed from the southwest looks like a rough-hewn statue of a nun kneeling before the altar; hence, the name "Kneeling Nun" by which the landmark, prominent for many miles, is known.

Passing in a generally northerly direction from the point of Santa Rita Mountain is a ridge of lower altitude, exposing the underlying Cretaceous rocks which form the northeast boundary. Further toward the north the Permian and the limestones of the Upper Pennsylvania, with a dip toward the basin, form the northeast to north boundary. From the north to the northwest the monocline exposes the Permian and upper limestones, dipping toward the basin, which form the high ridge between Santa Rita and Hanover Creek. The main spur from this ridge at a lower elevation extending to the south and turning southeasterly makes a less prominent ridge around the basin on its northwesterly and southwesterly rim, showing exposures of Permian limestone, Cretaceous sediments, and quartz diorite porphyry.

Santa Rita Creek flows from the northeast into the basin and cuts through the southwesterly boundary. The Sante Fe Railroad enters the basin through the creek gap in the southwestern rim. The mine offices and business

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section are located on the granodiorite core, while the greater part of the residential section lies at a higher elevation and to the north, northeast, and east of the pit area. See Figure 4.

The ore is found in the granodiorite and in the quartz diorite porphyry masses, as well as in the sediments themselves. The alterations of the rocks have been so complete as to make differentiation of the sediments from the igneous masses most difficult, especially in the proximity of and within the mineral-bearing areas. In the masses where silicification has taken place the ore is hard and generally tough. In the sericitized and kaolinized portion the ore is soft, containing much colloidal mineral.

The thickness of the overburden is extremely variable and ranges from nothing, over the northwestern and northeastern portions of the ore body and small areas in other sections, to about 150 feet over the southern portion.

A line drawn through the center of and connecting the Santa Rita open-pit ore bodies would outline an irregular ellipse with its axis trending about North 40° West. The central dome of granodiorite is eccentrically located in the ellipse, giving a width of ore of 400 feet on the northwest, 1,000 feet on the northeast, 2,000 feet on the southeast and 1,200 feet on the southwest. The length around the outside perimeter of the ore bodies is roughly 13,500 feet and the distance around the granodiorite core is about 7,500 feet (fig. 5).

The contours of the top and bottom of the ore are very irregular. In the main part of the ore body the thickness of the ore reaches 600 feet, feathering along the margins to 300 feet and less. None of the ore bodies have been delimited by drilling either vertically or horizontally. Structure studies indicate extensions of importance in both dimensions.

Malachite, azurite, chrysocolla, dioptase, cuprite and native copper are present in the upper zones of the ore body, especially along the more prominent faults and fractures. However, chalcocite and chalcocite-coated pyrite are the most important minerals, generally disseminated through the rocks, although a portion of the values are carried in small seams and stringers. Quartz, sericite, halloysite, and kaolin are common gangue minerals. Leaching of the upper zone accompanied by subsequent secondary enrichment has made the ore body one of economic importance. The grade of ore varies from 0.80 to 2.00 per cent and over, with an average grade for the entire remaining open-cut ore body, as now estimated, of 1.27 per cent copper.

METHOD OF PROSPECTING

Thousands of feet of shaft and underground workings, disconnected though they were, furnished preliminary information as to the presence of rather extensive low-grade copper deposits justifying a drilling campaign to develop definite tonnages.

Churn drills of the "spudder" type, capable of drilling to a depth of 500 feet were used on the first holes; later, churn drills with a range of 1,200 feet were used. The first drills were all actuated by steam power. At present, drill rigs powered by gasoline engines are in operation. In the

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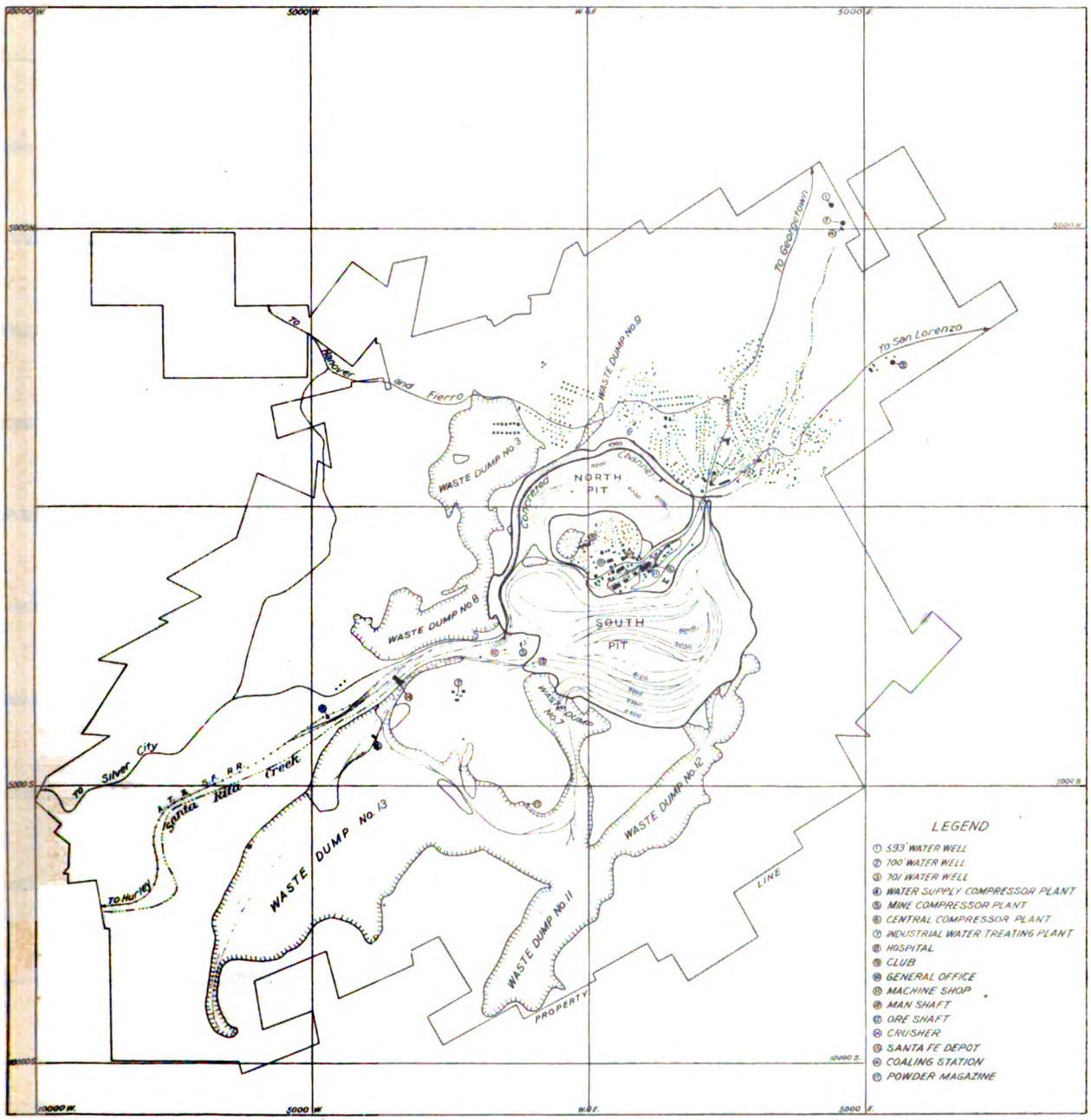

Figure 4.-Property map of Chino mines

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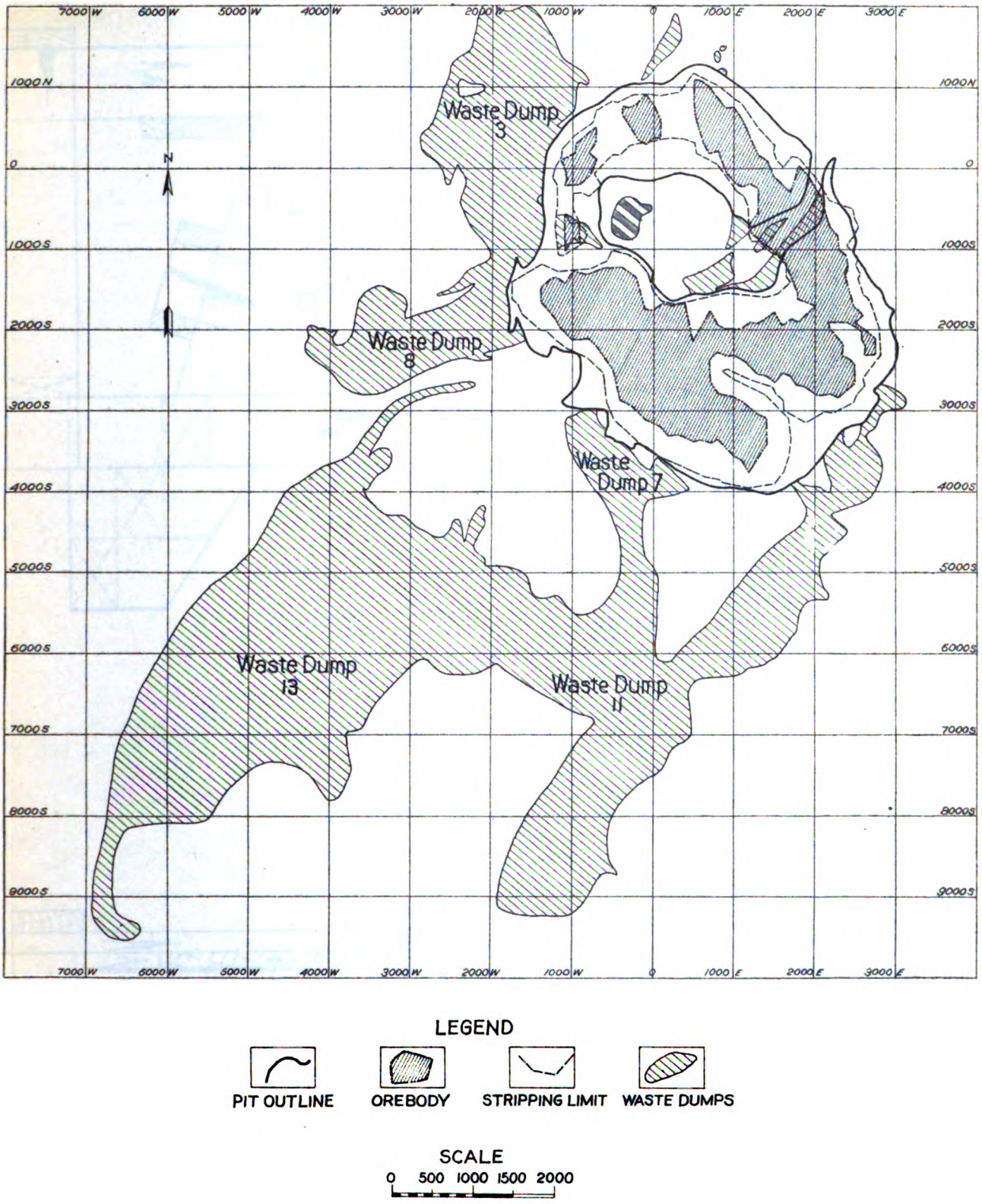

Figure 5.- Orebody, stripping limits, pits and waste dumps

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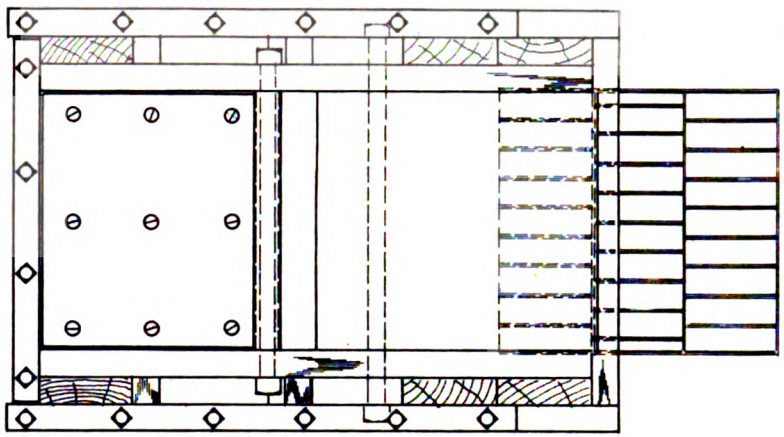
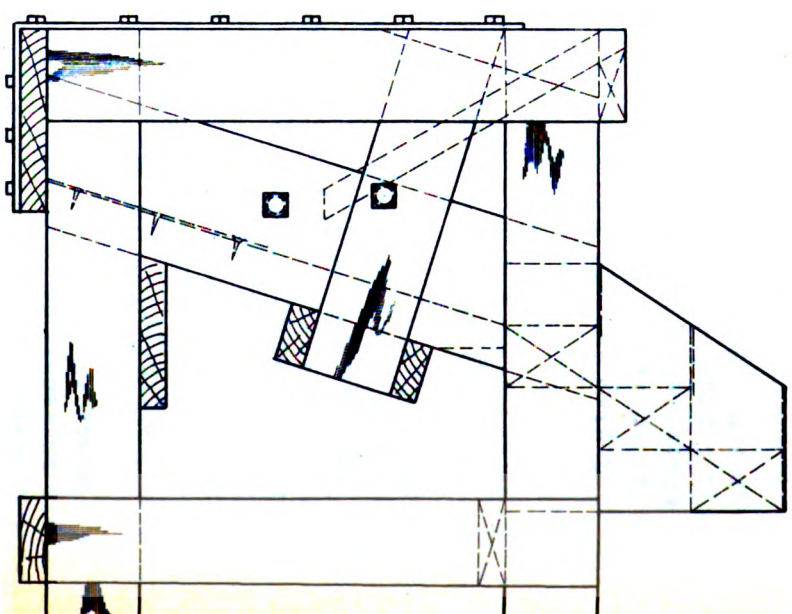


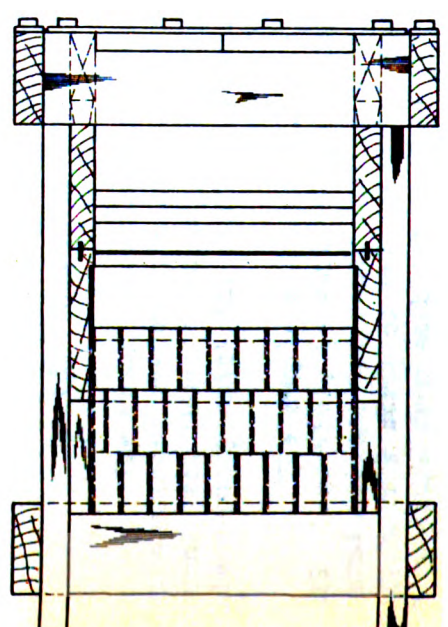
TOP VIEW

SCALE

0 1FT. 2FT. 3FT. 4FT



SIDE VIEW



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beginning holes were drilled at 100-foot centers on coordinate lines. This distance was increased to 200 feet, which gave dependable data for estimating tonnage and grade of ore.

The prospect holes drilled in later years have averaged about 900 feet in depth. They are usually started with a 13-inch bit and continued with this size until caving becomes serious. Casing is then inserted and drilling is continued with a small-diameter bit. In some instances tools as small as 3 inches in diameter are used. Underreaming is not used except as a last resort. The hole is cased when ore is first encountered, at any depth that caving is apparent or suspected, and again upon leaving the ore zone. Every precaution is taken to obtain an accurate and representative sample of such depth interval as was selected for the hole.

SAMPLING AND ESTIMATION OF TONNAGES AND VALUES

Sampling of Prospect Drill Holes

During the first half of the prospect drilling campaign samples were taken at intervals of 3 feet; however, samples at every 5 feet as taken now are proving satisfactory. All of the sludge from a 5-foot interval is bailed out and passed through a large Jones-type sampler equipped with splash boards and a mixing chamber. See Figure 6.

The sample is carefully split, one part being preserved in the wet state and the other part dried.

The wet sample is saved until assay returns are received on the dry part. Composites of wet samples in the ore bands are then made up and metallurgical tests are run on them.

The dry sample is divided up and used for the following purposes:

(a) Sample for mine assay office.
(b) Sample for mill assay office.
(c) Sample washed clean of sludge and dried for geological department.
(d) Remainder of pulp retained for possible future use.

The assay laboratory at the mine determines the copper by the potassium iodide method, whereas the mill assay office runs them electrolytically. The returns from these two assays usually check within very close limits, but should a discrepancy exist the samples are rerun by each office.

The sample washed clean of sludge is used in two ways; one part, in a small glass vial, is sent to the geological department where it serves for rock determinations; the other part is used on the sand board. The sand board is a long, narrow strip of wood on which the particles of drill cuttings are glued; the cuttings from each interval of the hole are put on to scale. This is kept up to date as the hole is drilled. The bottle samples are kept in specially constructed racks and are thus always available for future reference.

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Daily reports are made by the drill crews as well as the assay office. Records are kept posted daily and upon the completion of a hole a log is constructed by the geological department (fig.7).

Following is a copy of the progress record for two holes recently drilled, one with a steam rig and the other with a machine equipped with a gasoline engine.

Progress report of two holes drilled by different power

	Drill hole 824	Drill hole 825
	Steam	Gasoline
Elapsed time (days)	66	45
Shifts worked, total	112	83
Shifts moving and setting up	7	3.66
Shifts drilling	102.08	76.42
Shifts casing	2.92	2.92
Footage drilled	945	965
Average progress per shift, total shifts	8.44	11.62
Average progress per shift, drilling	9.25	12.62

Following is a segregated cost statement for the same two holes:

Segregated cost statement for two holes drilled by different power

	Drill hole 824		Drill hole 325	
	Steam		Gasoline	
	Total cost	Cost per foot	Total cost	Cost per foot
Moving and setting up, labor	$ 159.51	$ 0.168	$ 84.65	$ 0.088
Drilling, labor	2,326.16	2.463	1,764.26	1.828
Casing, labor	66.46	0.070	67.34	0.070
Fuel	421.44	0.446	221.35	0.229
Supplies	67.30	0.071	42.06	0.044
Sampling and assaying	475.73	0.503	469.79	0.487
Totals	$ 3,516.60	$ 3.721	$ 2,649.45	$ 2.746

Permanent records are made of each drill hole showing the assay of each interval sampled, color of sludge, principal minerals, and the rock formation as reported by the geological department. Accompanying each set of sheets for a finished hole is a summary sheet showing depth and average assay of capping or ore, and of any waste bands. The elevations of the top and bottom of each band of ore or waste are shown. Cross-section sheets are also developed on a scale sufficiently large to record the individual assays.

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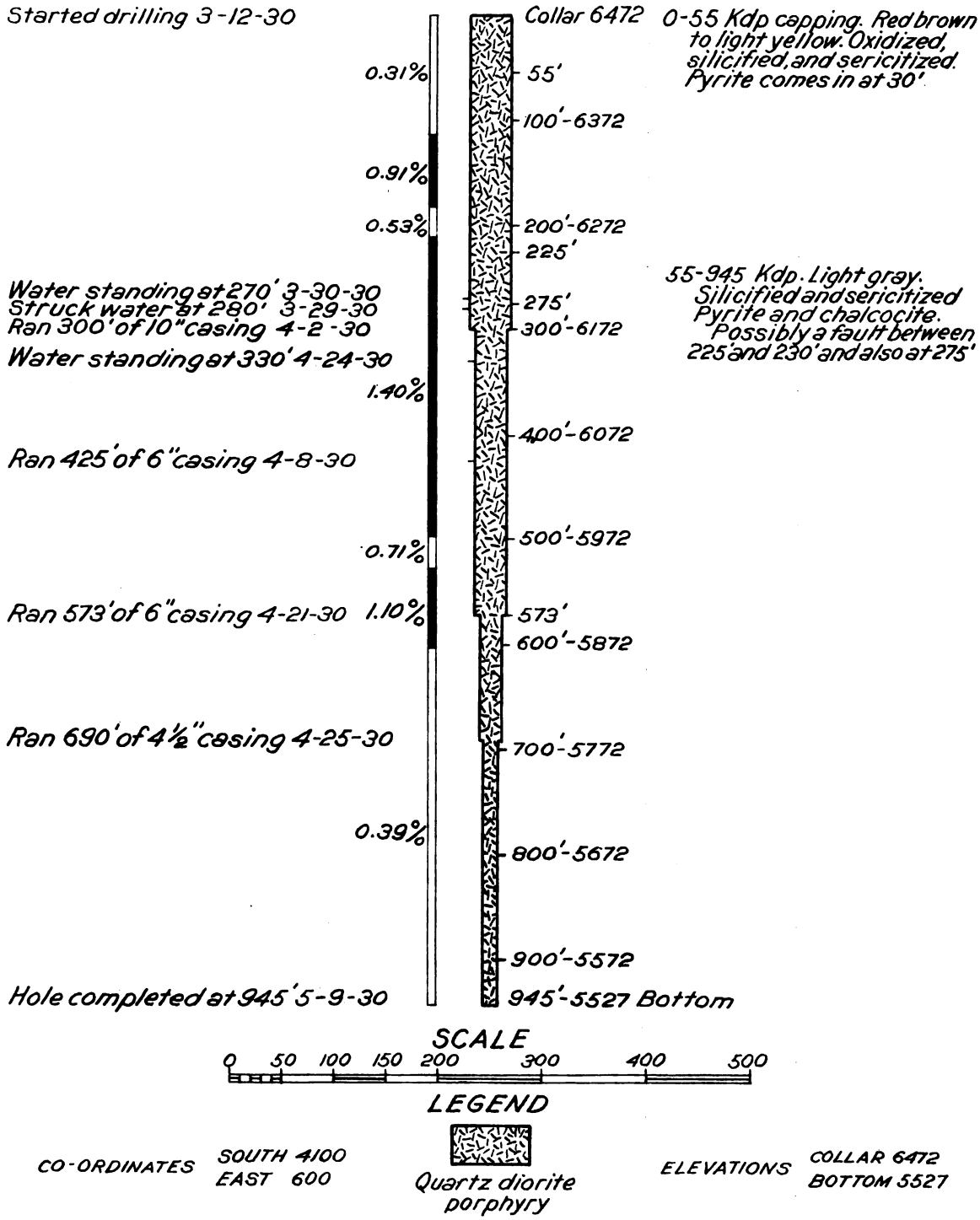

Figure 7.- Churn drill hole No. 824

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Ore Reserves

A point of origin for a system of coordinates was located on the central core of Cretaceous granodiorite previously mentioned, for the purpose of ready reference of engineering field data regarding topography, tracks, buildings, etc., and all surface or underground workings. Any point, such as a drill hole location, is expressed as being so many feet north or south and so many feet east or west of the point of origin. In order to preserve this system of coordinates a number of permanent triangulation stations have been established on prominent points entirely outside the present known area of power-shovel operations. United States Geological Survey sea datum elevations have been adopted.

As a preliminary to the calculation of ore reserves sections are developed along the meridian lines and at right angles thereto. On these sections surface elevations, drill holes, and any underground workings with assay values are shown, together with such other data as may be necessary. From the data and experience gained during past operations, drill-hole assays are grouped into bands of ore and of waste and outlined on the sections. Maps are developed from these sections showing the ore intercepts for each 50-foot bench, together with such other data as may be necessary to the calculation.

Having thus outlined the ore zones the selected pit slopes are drawn on the sections in such a manner as to include the maximum amount of ore with a minimum amount of waste or stripping. The ratio of ore to stripping should not average much above 1 ton of ore to 0.75 cubic yard of stripping (1 ton of ore to $1\frac{1}{2}$ tons of stripping) for the whole ore body. Other factors, the area of the ore at the lower levels, the length of haul, and practical grades and curvature for service tracks, determine the depth to which the power shovels can be economically operated and must be considered; they are all given due consideration in placing the slopes of the pit on the cross sections and in outlining them on the maps.

In practice it has been found that except in two comparatively small areas a slope of 1 : 1 (45°) from the intersection at the surface to the bottom of the pit is safe. The bank on any individual bench will stand nearly vertical for a period of years. No material trouble arises from slides if one shovel cut about 25 feet in width is left in the slope. The bench serves to hold the spalls resulting from weathering.

A calculation is made, using average and areas on the sections, to determine the total volume of ore and waste to be moved. A separate calculation is made of the ore. From the volumes are calculated the tons and average grade of the ore and the cubic yards of waste.

The general outline of the shovel excavations as shown on the sections and bench plats may be modified from time to time to accommodate changes in the transportation system that will insure the most economical disposal of material to be removed.

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All ore lying below the bottom of the power-shovel pit and outside of the slopes and all ore where the stripping ratio is much greater than stated above is considered as underground ore and so treated in any calculation of reserves. The completion of any additional development necessitates a calculation of reserves from the new data. A realignment of pit and underground ores usually results.

It has been found that an average bank or bench height of 50 feet is most practical for all classes of material handled from the pit. Therefore, as an aid in following the extraction it has been found practical to estimate the power-shovel ore and waste by benches. Each bench, or bank, is designated by its elevation above sea level. The proper assay value is assigned to each 50-foot bench for its length along a section.

The ore and waste zones on the benches as developed by actual mining are plotted daily by grouping of assays. The recorded weighed tonnages with the average assay for the quarter are used by the engineering department for production reports. These also serve as a basis for calculations of the ore mined in the calculated ore zones and that mined in areas not included in the ore-reserve estimate.

During the period since the last detailed ore calculation was made it has been found that the actual ore removed has been 120 per cent of the estimated ore tonnage and has contained 123 per cent of the expected copper content. The 20 per cent additional ore has come from within the stripping slope where it has been extended outside the drilled area of the calculated ore zones, and from beyond the estimated stripping slope where excavations have encountered ore.

Because of certain structural details it has been found convenient in making the calculations of power-shovel ore reserves to recognize four distinct ore bodies. Those four, the North, Northwest, Southwest and South ore bodies are the result of the consolidation of what in the early days were 10 separate groupings. As further drilling is completed, some of these four designations may be dropped. Aside from the designations that are in use for pit ores there are two that apply only to underground-ore calculations.

The final record of the bench calculations of one of the ore bodies illustrating the method of segregation of ore and waste by benches follows:

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Final record of the bench calculations of an ore body in the Santa Rita district

Bench	Total cubic yards	Waste, cubic yards	Ore, tons	Per cent copper	Ratio, tons waste to tons ore
6500	33,072	33,072	---	---	---
6450	379,912	379,912	---	---	---
6400	780,777	780,777	---	---	---
6350	1,010,846	909,945	201,802	0.98	1/ 20.85
6300	1,371,066	1,066,124	609,885	0.88	3.50
6250	2,377,641	1,656,623	1,442,037	1.35	2.30
6200	4,229,382	2,693,526	3,071,713	1.21	1.75
6150	5,074,787	2,352,450	5,444,674	1.27	0.86
6100	4,933,851	1,325,098	7,217,506	1.30	0.37
6050	4,594,381	1,114,022	6,960,717	1.31	0.32
6000	3,602,748	753,248	5,693,000	1.30	0.27
5950	2,749,777	609,207	4,281,141	1.37	0.28
5900	2,159,759	465,926	3,387,667	1.37	0.28
5850	1,295,467	136,278	2,318,378	1.19	0.12
Totals or averages	34,593,466	14,279,208	40,628,520	1.29	0.70

1/ This bench charged with all waste from three upper benches.

Measurements of Material Removed

Present practice is to obtain engineering data by actual field surveys at the end of every quarter, plot the extension of the benches on the cross sections and bench plats, and calculate the solid yards of material removed for that quarter. A daily record is kept of the cubic yards contained in cars of ore or waste removed from each one of the benches. At necessary intervals during the three-month period the location of each shovel is noted, so that the zones of ore and waste which the shovel has removed can be definitely recorded. The assay of the daily composite sample taken from each car is applied to the ore. Ore and waste loadings are sampled separately, greater care being taken and more frequent samples obtained of the ore than of the waste. The field sampling of the ore shipped to the mill is checked daily with the automatic sample of mill heads and if corrections are necessary they are applied to the ore mined from the various bench zones, in proportion to the pounds of copper from each zone as shown by the daily field samples.

The periodic engineering calculations together with operating records provide the necessary basis for proportioning between the various benches the ore and waste mined inside as well as the ore and waste removed outside the calculated ore zones but inside the stripping slope. Waste and ore moved outside the slope can also be apportioned. These data also serve the purpose of checking the factor used during the period for converting car cubic yards into solid cubic yards in place. If it is found that the factor has changed, the calculations for the third month of the period receive the necessary correction and the new figure is applied to the succeeding quarter. This factor fluctuates between

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76 and 78 per cent. The total car cubic yards multiplied by this factor gives the total cubic yards (solid) in place. A standard can thus be made of the loading of cars by the shovel runners and their efficiency checked.

In making the final record (Table 1) for an ore body the total cubic yards, in the solid, moved for each bench is recorded in the second column. After making the separation of ore and waste, the cubic yards of waste and its average assay are recorded in columns 3 and 4 , while columns 5 and 6 show the total tons of ore and its percentage of copper mined from each bench. The sum of the bench tonnages represents the total amount of ore that was shipped during the quarter. Columns 7, 8, and 9 under "Material Removed from Outside of Ore Reserves" show the total cubic yards moved, the tons of ore, and the percentage of copper it contains. Under columns 10 and 11 are shown the ore expected in the calculated ore zones and its corresponding assay value. In columns 12 and 13 are shown ore actually mined in the calculated ore zones and the assay value as determined by mine sampling revised to check with the mill samples.

"Total ore," shown in column 5, is the sum of the ore shown in column 7 plus the ore shown in column 12 and is made up of the ore actually mined in the calculated ore zone and the tons of ore mined outside the ore reserves. To subtract this total from the original ore reserves calculated would give an incorrect figure. The tons of ore expected from the ore zones is shown in column 10. These are the figures that are deducted from the tons of ore shown in the original calculation and will give a correct figure of the ore that will remain in that ore body, together with the correct assay value. The new figures for the ore body as to tons of ore and grade, as well as cubic yards of waste still to be moved, are not affected by the tonnage of ore or cubic yards of waste outside of the original estimate.

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Below is a copy of one of these quarterly statements which applies to the ore body for which the total calculation was shown previously.

Table 1. - Quarterly statement of production.

1	2	3	4	5	6	7	8	9	10	11	12	13
Bench	Total cubic yards	Waste, cubic yards	Copper, per cent	Total ore		Material outside of ore reserve			Expected in ore zone		Mined in calculated ore zone	
				Tons	Copper, per cent	Total cubic yards	Ore tons	Copper, per cent	Tons	Copper, per cent	Tons	Copper, per cent
6500	784	784	0.10	---	---	---	--	---	--	--	--	--
6450	37,430	37,430	0.11	---	---	34,467	--	---	--	--	--	--
6400	80,448	80,448	0.22	---	---	27,155	--	---	--	--	--	--
6350	57,968	51,137	0.20	13,662	0.86	12,941	13,100	0.86	562	0.84	562	1.03
6300	68,133	54,235	0.21	27,795	1.05	18,648	14,803	1.05	12,992	0.87	12,992	1.05
6250	68,306	11,303	0.26	114,006	1.30	22,252	84,502	1.31	29,504	1.01	29,504	1.26
6200	212,652	72,320	0.41	280,664	1.49	27,244	20,798	2.10	259,866	1.24	259,866	1.45
6150	357,332	191,674	0.43	331,315	1.28	30,711	50,575	1.00	280,740	1.35	280,740	1.34
6100	25,007	12,859	0.26	24,296	1.72	5,396	7,340	1.72	16,956	1.31	16,956	1.72
Total	908,060	512,190	0.32	791,738	1.36	178,814	191,118	1.28	600,620	1.27	600,620	1.38

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Sampling

A sampling department checks ore and waste delivery and provides data concerning the daily operation. It is directly under the supervision of the mine engineer. Individual samplers are in charge of a field foreman, who places trained samplers at each shovel working during a shift. Samples are taken from each car of ore of a train as loaded and are delivered to the sample room at definite intervals during a shift; these represent a definite number of cubic yards of broken ore. The average daily assay of the ore loaded by each shovel in ore is determined from the composite sample made from these pulps.

When a shovel is loading material that may be either ore or waste, delivery to the primary ore crushing plant or waste dump is directed by the sampler at the shovel. In order to check the judgment of the sampler, more frequent samples are taken and immediately delivered to the assay office for determination. The sampler's designation of ore or waste is checked as promptly as possible and, when necessary, the destination of the material is corrected by the assay office notifying the sampler and the main dispatching yards. The time needed to check the sample, designate it as ore or waste, and inform the dispatcher at the main yards does not exceed 45 minutes. The amount of waste that might be mixed with the ore and ore that might be loaded out as waste is but a minor quantity. Through daily practice the judgment of the sampler becomes remarkably accurate.

MINING METHODS

General

During more than a century before the acquisition of the Santa Rita mines by the Chino company thousands of feet of shafts and other underground workings were driven along high-grade stringers. This work was done principally where surface outcrops were most favorable. Many crosscuts were driven into the main rock mass, disclosing copper minerals that in these later years have become of commercial value. On the north and east portions of what is now the power-shovel pits were ore outcrops which with the underground openings permitted an estimate of about 13,000,000 tons of proved ore, as well as about 18,000,000 tons of possible ore. Preliminary drilling immediately increased the proved tonnage. It became apparent early in the development of the property that power-shovel mining methods should be selected. This decision was the more readily accepted as the staff of the Utah Copper Co., where steam-shovel operations had been underway for about three years, was directly connected with the Chino operations.

The Atchison, Topeka & Santa Fe Railway provided an adequate transportation system for the moving of ore to any point selected for the mill, as well as to the smelter at El Paso, Tex. It was possible to select points for beginning the removal of ore by making a connection of the industrial track for shovel service with the railway terminal facilities.

The topography of the surface was adapted to the construction of trackage at reasonable grades and cost for removal of the overburden to areas on the property where open-pit ore bodies were not likely to be developed. The

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outcrops of some of the ore bodies and the surface topography made it possible to begin operations, with but little preliminary preparation at these points. It was necessary to stock-pile some ore before the completion of the mill at Hurley, a point on the Atchison, Topeka & Santa Fe Railway 10 miles from Santa Rita.

Previous to July 1, 1910, two steam shovels had been ordered with the necessary locomotives, cars, and track material. The first shovel was received and erected at the old railway station site and placed in service on September 23, 1910. It started making the preliminary cut from the end of the railroad around to the north side of the present pit in the area then known as the Romero ore body. By September 30, 1910, the second shovel had been received, erected, and placed in service in making a grade to connect with a waste-dump area. By December 1, 1910, the first two shovels were operating in the actual removal of stripping from the 6370 and 6340 levels of the Romero (North) ore body. A third shovel was doing preparatory work over the Chino-Yosemite (Northwest) ore body on the 6330 level. Before the end of 1910 a fourth shovel had been received, erected, and placed in service in making a track connection from a lower point on the Santa Fe to the Romero ore body.

The initial stripping operations were started on the Hearst, or South ore body, in 1911, thus establishing the main track connections to the ore bodies most fully developed at that time. The first ore was shipped to the mill in September, 1911.

Owing to topographic conditions, the first benches were 30 feet high, but as operations continued it developed that a 50-foot bench was the most economical height for the class of material to be handled at Chino. With the 50-foot bank more material is moved with less delays for moving shovels. Moreover, less drilling, blasting, and track work are required per cubic yard loaded than with the 30-foot bank.

Working in banks higher than 50-feet increases the hazard of missed holes, sliding banks, and rolling rocks, all resulting in unwarranted delays.

As development drilling continued to show extensions of all ore bodies, it became evident soon after the beginning of shovel operations, that the railroad station, located in a flat northwest of the central granodiorite core, and several thousand feet of tracks would have to be moved. Satisfactory arrangements were made with the railway company for the moving of tracks and station to a point south of the central core, north of the then known limits of the south ore body, in the immediate neighborhood of the business section. This change of trackage was accomplished by the end of 1911.

It had been recognized that eventually the main drainage through the Santa Rita basin would have to be diverted; the preliminary estimate, before much drilling had been done, placed the cost of this work at $150,000. As development drilling progressed it became evident that the diversion of Santa Rita Creek would involve a much greater change than had been originally considered. In 1916 another estimate was made placing the cost of removal at $750,000. Further

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development drilling demonstrated, moreover, that the railroad tracks and station would have to be moved again. Late in 1924 final studies were made and late in 1925 construction work was begun for diverting Santa Rita Creek from its original location around the perimeter of the north and northwest ore bodies to a connection with the old bed of the stream not far above the primary crushing plant at Earnest Siding (see 14, fig. 4) on the Atchison, Topeka, and Santa Fe Railway. The yard tracks and station were moved from their location in Santa Rita proper to a point below the primary crushing plant (see 15, fig. 4).

The flume for the diversion of Santa Rita Creek is constructed of reinforced concrete and has a total depth of 7 feet and a width of 19 feet 6 inches at the top by 18 feet 6 inches at the bottom. The flume itself was cast in an excavation made on a bench outside the stripping slope of the north and northwest ore bodies. The flume is approximately 7,500 feet long and is capable of carrying 5,300 cubic feet per second, approximately twice the observed maximum run-off from cloud-bursts. A detailed description of the construction of this flume, written by C. F. Ames, mechanical engineer of Chino mines, was read and published for distribution at the meeting of the New Mexico Chapter of the American Mining Congress held at Santa Rita, N. Mex., on July 25 and 26, 1927, the date upon which the final section was cast and the concreted channel was ready to receive any flood waters coming down Santa Rita Creek. The cost of the extra bench, flume construction, and removal of the railroad tracks, was approximately $900,000, but there was released for mining by power shovels approximately 12,000,000 tons of ore, besides a considerable tonnage already stripped which was made more easily available.

As the early shovel operations progressed it was found that the materia to be moved from the principal areas was exceedingly hard and tough, breaking in large masses along major fracture planes. Block-hole blasting and bulldozing were necessary, causing excessive shovel delays. This condition added considerably to the cost of loading ore. Moreover, it was impossible to break all the material to the minus 12-inch size necessary for rapid unloading at the mill bins The number of men required for unloading the ore at the mill bins and the numerous delays caused by large pieces in the railroad cars made some other arrangement necessary.

In 1913 the design for a primary crushing plant was finished and construction commenced, the plant being completed and put in operation in August, 1914. This crushing plant was equipped with a 48 by 60 inch jaw crusher set at 8 inches. Ahead of the crusher were heavy grizzly bars with 8-inch openings upon which the air-dump cars from the pits could deliver their load. The undersize from the grizzly passed into a bin and the oversize to the jaw crusher. The discharge of the crusher and the undersize were delivered into railroad loading bins by a pan conveyor.

The cost of the primary crushing and of loading the railroad cars from the bins was half of the cost of bringing the material down to proper size by bulldozing and blockholing when loading directly into the cars; a number of men were released and delays eliminated at the mill ore bins.

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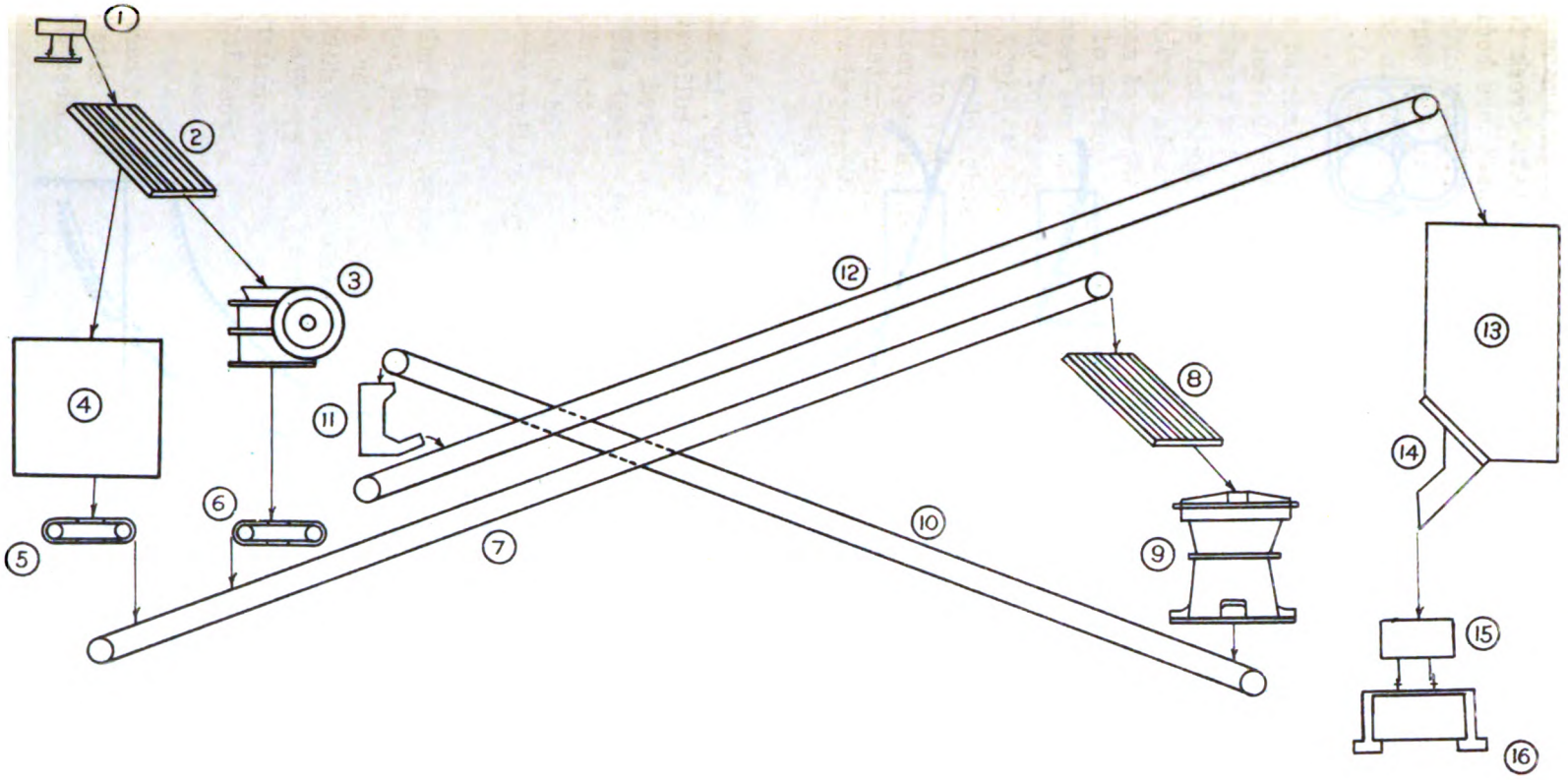

MARK	No.		MARK	No.	
1		20 or 30 cu.yd capacity side dump cars	9	2	#8 Gyratory crushers - both operating
2	1	Grizzly - 8" opening	10	1	60" belt conveyor
3	1	84" x 66" jaw crusher	11	1	Two way splitter
4	1	Undersize bin	12	2	42" belt conveyors - both operating
5	2	Undersize bin feeders - both operating	13	1	Storage bin
6	2	Crushed product feeders - both operating	14		Pneumatic operated gates
7	2	42" belt conveyors - both operating	15		55 ton capacity ore cars to Hurley Mill
8	2	Grizzlies - 3" opening	16	1	Railroad track scale

Figure 8.-Flow sheet of primary crushing department

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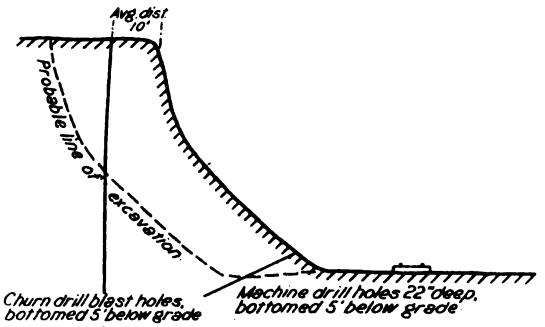

Avg. dist. 10'
Probable line of excavation
Churn drill blast holes, bottomed 5' below grade
Machine drill holes 22" deep, bottomed 5' below grade

Figure 9.- Usual drilling procedure

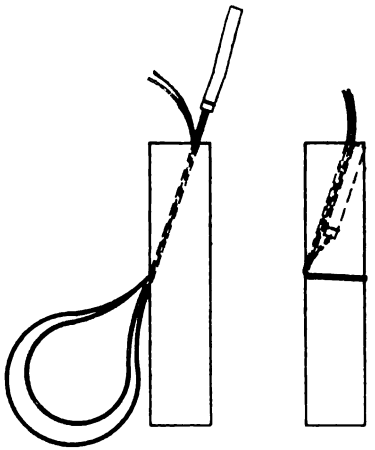


Figure 11.- Primers for use with electric blasting caps

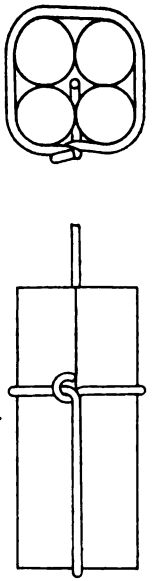
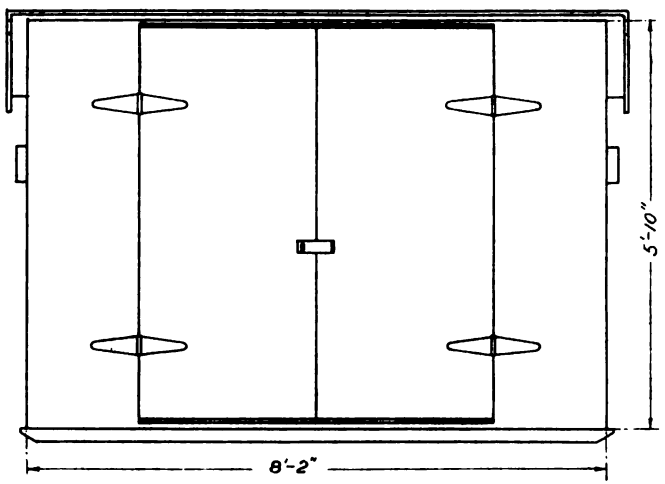


Figure 12.- Primers for bulldozing



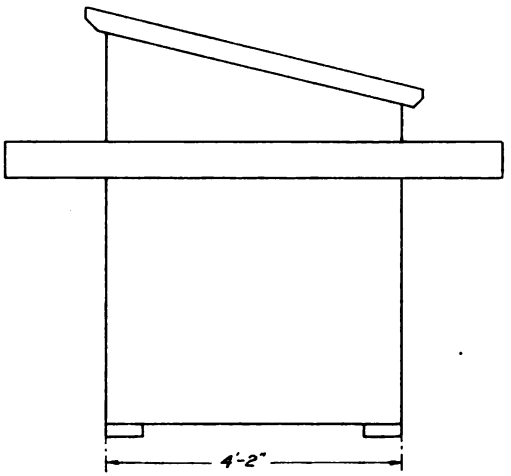


Figure 10.- Portable explosives magazine

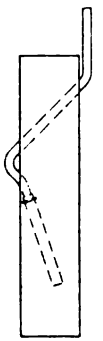


Figure 13.- Primers for use with cordeau

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As the capacity of the mill at Hurley was increased it was found necessary, in 1916, to place another primary-crushing plant at the mine. The design of the No. 2 primary-crushing plant was in general the same as that of the No. 1, except that an 84 by 66 inch jaw crusher was installed, thus increasing the size of the material that could be handled, as well as increasing the capacity. This second unit was completed and put in operation in May, 1917.

These two plants as originally installed delivered an 8-inch product into the Santa Fe ore cars for transportation to the mill at Hurley. In 1930 two No. 8 gyratory crushers were added to Plant No. 2, with the necessary grizzlies, feeders, conveyors, etc., as shown in Figure 8. The product now delivered for transportation to Hurley is approximately minus 4 inches in size. This has brought about further economies at the mill bins, both in labor and in lessened delays and repairs.

The crushing plants are located below the main delivery tracks from the pit. Ore trains discharge into the plant; waste trains continue to the dump. Mixed trains of ore and waste are thus handled conveniently. The coaling station is located just beyond the crushing plant on one of the main waste dumps where all trains are coaled and watered enroute (see 16, fig. 4).

Drilling

The drilling procedure varies slightly, due to the different character of rock encountered, though in general the system illustrated in Figure 9 is used. Changes in local conditions naturally modify practices. The blasting methods in use call for three different types of drills. For the primary blasting both machine and churn drills are used. The former are compressed-air hammer drills and the latter are churn drills operated by gasoline engines. For drilling boulders which have to be broken before loading by the shovels, light jackhammers are used.

During the early years of operation, piston air drills were used for the holes, but these were not entirely satisfactory, however, because owing to the heavy blasting the ground tended to ravel and fitcher, causing delay in finishing the holes. The earlier hammer drills were tried but their weak rotating mechanisms caused trouble. The latest types of hammer drills with more powerful rotation are satisfactory, and the piston drills have been discarded.

The air pressure at the drills varies with the distance from the compressors or air receivers. As far as practicable a minimum of 85 pounds pressure at the drills is maintained. Where the distance from the compressor mains is great, larger branch lines are used with receivers placed at convenient points, thus in a measure avoiding too great a drop in line pressure. A pressure of between 110 and 120 pounds is maintained at the compressors.

One and one-quarter inch hollow round lugged steel is used in drilling toe holes. These holes are started with a 3-$\frac{3}{4}$-inch bit and bottomed with 1-3/8-inch bits. Casing is seldom needed in toe holes. When necessary, because of the shattered condition of the bank, a short length of pipe is placed in the collar of the hole and removed if possible before the hole is loaded.

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Toe holes are ordinarily drilled to a depth of 22 feet. The depth of block holes for breaking boulders varies with the size of the boulder to be broken.

The drilling speed for toe holes varies with the softness of the ground. In a hard cap rock the average drilling speed if 5 feet per hour. In the softer rock the average drilling speed if 7-½ feet per hour. The average drilling speed based on total time for the entire pit is 5.8 feet per hour. For actual drilling speed the above figures should be increased 10 per cent to allow for the time of making a new set-up, connecting and moving air lines, etc.

The location and spacing of holes depends largely upon the ground, though a general plan is followed as far as possible. The toe holes, where conditions are normal, are spaced about 15 feet apart. The height of bank or depth of hole has no bearing on the spacing of toe holes. In places this spacing is not strictly adhered to, as when a hard point projects from the bank or high bottom is left by previous shots. In such cases a toe hole is started in the hard point, or bottom, irrespective of distance. All toe holes are intended to bottom 5 feet below grade, as shown in Figure 8.

The air-drill crew consists of a machineman and helper. The drill crews do their own shoveling preparatory to setting up the drills, except where an unusually large amount of rock has to be moved. In such cases a mucker is employed to clean out for set-ups. The drill crews maintain their own air lines.

The churn drills are of two sizes, the smaller size being equipped with 4-½-inch tools and the larger size with 6-inch tools. These drills are powered with gasoline engines and are self-propelling. In moving churn drills to distant points in the pits, or from one bench to another, the drills are loaded onto a flat car by a locomotive crane, moved to the new location by a switch engine, and unloaded by the crane.

Due to the shattered condition of the surface rock, a short piece of casing is always used in the collar of the hole, and is removed when the hole is completed, except in rare cases where it is apparent that the collar of the hole would cave were the casing removed. Even in such instances the casing is generally recovered after blasting.

Churn-drill holes are normally spaced from 20 to 30 feet apart, although this spacing may be varied according to height of bank and depth of hole. Where it is intended to break bottom with churn-drill holes, they are drilled 5 feet below grade. Where it is only necessary to trim the bank, churn-drill holes are generally sunk to a depth of 30 feet in a 50-foot bank and are called "slope holes." The spacing of "slope holes" is approximately the same as for "grade holes." All churn-drill holes are started about 10 feet from the crest of the bank. In places where any deviation from regular spacing is required, location of the holes is left to the judgment of the powder foreman.

The crew on the churn drills consists of a driller and helper. A "road gang" consisting of laborers and a boss does the necessary road building on the different benches for all the churn drills.

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Table 2 gives some data on total churn drilling for one month.

The sharpening of drill steel for both the machine and churn drills is done in a central shop. Standard drill-sharpeners are used for the machine steel and a specially built sharpener is employed for all churn-drill bits. All steel for both air machines or churn drills needing sharpening or repairs is delivered by hand to the nearest points of the track on benches in the pits. A switch engine with a specially designed flat car makes collection at least once a day of the dulled steel, delivering at the same time the sharpened steel from the shop.

Table 2. - Churn-drilling data for May, 1930

Drill number	Total hours worked	Total feet drilled	Feet drilled per hour worked	Time drilling, per cent	Time delay, per cent	Time repair, per cent	Feet per hour drilled	Cost per foot drilled
9	68	235	3.46	96	4	0	3.61	$ 1.68
12	112	820	7.32	97	3	0	7.53	0.32
15	139	702	5.05	100	0	0	5.05	0.62
17	351	1,537	4.38	89	10	1	4.92	0.63
18	152	604	4.00	93	0	7	4.30	0.93
19	128	895	6.99	99	1	0	7.04	0.43
20	264	1,395	5.28	91	5	4	5.77	0.55
21	72	386	5.36	79	21	0	6.79	0.66
22	208	898	4.32	92	8	0	4.69	0.66
23	229	1,580	6.90	96	4	0	7.18	0.41
24	367	1,775	4.84	97	2	1	4.99	0.69
25	264	1,177	4.46	98	2	0	4.55	0.75
Totals and averages	2354	12,004	5.10	94	5	1	5.41	$ 0.62

Blasting

Explosives are unloaded from railroad cars directly into an underground powder magazine with a capacity of five carloads, located on a spur from a dump track (see 17 on fig.4). From the main storage magazine, explosives are delivered to wood-lined steel powder boxes (see fig. 10) placed at convenient points on the benches. These boxes have a capacity of 2 tons, or of eighty 50-pound boxes of explosives. These bench magazines are light enough to be moved by hand when necessary. Caps and fuse are stored on the benches in small wood-lined steel boxes located at least 100 feet from the bench powder magazines.

After much experimenting the following explosives have been adopted as standard: Apache Quarry Special, Nos. 1 and 4; Apache Standard (ammonia) and gelatin dynamites, of 40 and 60 per cent strengths and in 1¼ by 8 inch cartridges; and Herco standard black blasting powder.

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The Quarry Specials are bulk powders, packed in 12½ pound bags, and are used in blasting all churn-drill holes except those which are exceedingly wet. The No. 1, of approximately 35 per cent strength, is used in soft ground; the No. 4, corresponding roughly to 50 per cent strength, is used in the harder ground. In very wet churn-drill holes gelatin explosives are used. Four by eight inch cartridges have been tried for blasting churn-drill holes; because of the rough surfaces of most of the holes, however, difficulty was encountered in getting the cartridges to the bottom, and the use of large cartridges was discontinued.

The standard 40 per cent strength dynamite is used in toe holes in soft dry ground, and the 60 per cent in hard dry ground. The gelatin powders are used in all wet holes, the 40 per cent strength in soft and the 60 in hard ground.

Black powder is used for blasting churn-drill holes in soil and in very soft, kaolinized materials.

No. 8 electric blasting caps are used in all toe holes. Countered cordeau Bickford is used for detonating dry or only slightly wet churn-drill hole charges. It has been found that this fuse can stand some moisture if the end is tightly crimped. Cordeau Bickford is used wherever possible, as better over-all results are obtained. The cordeau is detonated with No. 8 electric blasting caps, fired with a battery. In churn-drill holes containing much water it is necessary to use No. 8 electric blasting caps. No. 8 blasting caps and fuse are used for secondary blasting consisting of plug and bulldozing shots.

Primers are made as shown in Figure 11 when electric blasting caps are used, and as in Figure 12 when cordeau is used. In the latter case the weight of the cartridges is an aid in lowering the cordeau Bickford in the hole. In all holes two primers are used, the first being at the bottom and the second in the charge about two-thirds of the way from the bottom.

For blasting plugs in boulders the cap is simply placed in the hole so that there will be powder both below and above it. Primers for bulldozing or "dobie" blasting are made as in Figure 13.

The springing of all holes is done with stick powder, detonated with electric blasting caps. The amount of charge and the number of times it is necessary to spring the hole depend upon the character of the ground. No set rule for springing is in effect, this being left to the judgment of the powder men, who become very proficient through experience gained in working hard and soft banks. When, owing to hardness of ground, it is necessary to spring more than once, a small charge consisting of from 15 to 20 sticks of 40 per cent strength explosive is used in the first springing. Subsequent charges are increased until it is known that a sufficiently large chamber has been made to hold the required charge for the main blast. Water is used for stemming in all springing shots. The quantity poured in serves as a rough measure of the size of the chamber; the water also cools the hole and less solid material is thrown out of the hole by the springing blast than when sand stemming is used.

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For determining the condition of churn-drill holes after springing, a gage is used which consists of a wooden block 4 inches in diameter and 36 inches long weighted at the bottom with 3 to 4 inches of babbitt. This gage, which is fastened to a soft rope, is lowered into the hole to measure the size of the chamber; the temperature of the hole is roughly determined by feeling the babbitt when the gage is withdrawn. In the toe holes a round tamping stick 22 feet in length and slightly smaller than the hole answers the same purpose. Except when large and repeated springing charges have been necessary it is seldom that much time is lost in waiting for a hole to cool before loading.

The churn-drill holes are loaded through a funnel, the bags being emptied directly into the hole. When stick powder is loaded into a churn-drill hole the powder is previously cut into three or four parts and loaded through the funnel. Not more than three men are allowed to load churn-drill holes; one man opens the boxes and the other two load. Dirt for stemming is first passed through a one-half inch screen.

The toe holes are loaded with a long tamping stick, one end of which is pointed. In loading a rough hole the point of the stick is pushed into a stick of powder, which is then placed in the hole and forced to the bottom. One man only is employed in loading toe holes. No stemming is used.

All firing of holes is done by hand batteries with a capacity of 100 caps each. As far as possible not more than 10 holes (20 caps) are fired at one time--a large factor of safety. Where two or more electric detonators are used or more than one hole is to be exploded, connections are made in series. Where cordeau Bickford is used, one electric exploder is placed on the trunk line for setting off the blast.

The size of the powder crews varies, but averages approximately two men per shovel on day shift and 1¼ men on afternoon shift, depending on the number of plugs or bulldozing shots necessary in addition to the toe and bank holes to be blasted. Aside from springing, the time of loading holes varies but will average about 5 minutes for a churn-drill hole and 15 minutes for a toe hole.

Including both primary and secondary blasting 6.04 tons of material are broken per pound of powder for all benches.

In certain parts of the pits a great deal of secondary blasting is necessary. Whenever possible the large boulders are drilled with jackhammers, though it is often necessary to break boulders with "dobies" when many are uncovered by the loading shovel. The secondary blasting consumption is 15 per cent of the total powder used for the upper benches of the southwest ore body, but averages 4 per cent for all banks.

Loading

The original equipment at Chino for power loading was the railroad-type Model 91 Marion steam shovel equipped with a 3½-yard dipper. When additional equipment was purchased in 1913 the Model 92 railroad type became the standard,

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and as repairs were necessary the old Model 91's were made to conform with the stronger Model 92. These shovels were all mounted on railroad trucks.

In December, 1923, the first power shovel was equipped with caterpillar tractors or crawlers. This resulted in a saving of about $8 per shift in labor and, owing to greater mobility of the tractor shovel, in increasing its loading capacity as figured over a year's service to 10.5 per cent more than that of the railroad truck shovel, thus effecting a net saving of 3.6 cents per cubic yard of material moved. The railroad-type steam shovels were gradually equipped with crawlers in place of the railroad trucks, except three which were reserved for special work in soft wet ground or on switchback tracks where continual moving from one level to another was required.

In May, 1925, the first Model 92 direct-current electric shovel equipped with caterpillar tractors was purchased and placed in operation; a second one was received and placed in operation in January, 1927. Beginning in 1925, whenever a steam-operated shovel needed general overhauling and extensive repairs of mechanical equipment it was dismantled, and the old, worn-out steam equipment was removed and replaced by the necessary electrical equipment. Up to the present time five of the original steam shovels have been repaired and provided with the electrical equipment of the standard type and with caterpillar tractors.

During one year of normal operations when the shifts operated by steam shovels and by electric shovels were nearly equal in number, it was found that the electrically operated shovel showed a saving of more than $70 per 8-hour shift of operation.

A Marion electric Model 350 shovel mounted on trucks and of the complete revolving type was purchased and placed in operation in July, 1924. This shovel is equipped with an 80-foot boom, 57-foot dipper sticks, and an 8-yard dipper; it loads with ease on a track 42 feet above its level and makes a thorough-cut 124 feet wide at the base. It was purchased for drop and thorough-cut work and has paid for itself many times since its installation.

In 1926 one Model 30-B Bucyrus diesel-engine shovel mounted on caterpillars was purchased for general utility work. This shovel will load and unload itself by its own power on a flat car and can, therefore, be readily moved from one point in the pit to another. It has been extremely useful as well as economical in making preliminary grades for standard shovels when a new bench is to be established. It was also most efficient in cutting the ditch for the construction of the Santa Rita Creek diversion flume.

The present shovel equipment is as follows:

One Marion Model 350 electric shovel with 8-yard dipper.
Seven Marion electric Model 92 shovels mounted on caterpillars.
Three Marion steam shovels, Model 92, mounted on caterpillars.
One Marion steam shovel, Model 92, mounted on railroad trucks.
One Model 30-B Bucyrus diesel-engine shovel mounted on caterpillars.

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The Model 92 shovels are equipped with 4-yard dippers; the Model 30-B Bucyrus with a dipper of 1¼ yards capacity.

Power for the operation of the electric shovels is transmitted from the power plant at Hurley to Santa Rita at 24,000 volts, 3-phase, 60-cycles. There it is transformed and delivered to the shovels as 4,000-volt alternating current.

Each electric shovel is equipped with a motor-generator set consisting of an alternating-current, synchronous motor and three direct-current generators all on a common shaft. One generator is for the two hoist motors, one for the crowd or digging motor and one for the swing motor. Direct current is supplied at 250 volts and operation is through Ward-Leonard control.

A pole line carries the transmission wires from a central transformer station to a switch house on each bench or level; from there it is transmitted through a four-conductor, No. 4, 49-strand, "Ozex"-type Tirex cable to each shovel. On the rear end of each shovel is attached a manually operated cable reel of 2,400 feet capacity by which the cable is paid out as the shovel advances and is rewound as the shovel retreats along the bench.

Power consumed by the standard Model 92 shovels is 0.75 kw. h. per cubic yard of material loaded; the Model 350 shovel uses 1.18 kw. h. per cubic yard.

Due to a lesser proportion of the harder cap rock being removed, together with the continued improvement in loading and transporting equipment and facilities, the cubic yards of material loaded per 8-hour shovel shift has gradually increased from 825 in 1923 to 1,280 at present. During the year 1929 the Model 350 electric shovel moved 1.7 times as many cubic yards per shift as the average moved by the Model 92 electric shovels.

The crew for a steam shovel equipped with railroad trucks consists of an engineer, a craneman, a fireman and six pitmen, whereas the crew of a standard electric shovel is an engineer, a craneman, an oiler and one pitman; the crew of the large electric shovel is an engineer, an oiler and three pitmen. One electrician on each shift takes care of all shovels.

Transportation

The entire mine operating efficiency depends on adequate transportation facilities. Therefore, it is essential that an ample number of locomotives and cars be provided, that sufficient passing tracks be properly located and that coaling and watering stations be placed so that train service causes a minimum delay at the shovels.

All tracks are standard gage (4 feet 8½ inches). In laying out the service lines no curves sharper than 20° are used and no grades over 2.5 per cent, compensated on curves, 0.4 per cent for each degree of curvature. Tracks in pits and on the dumps are laid with 75-pound and the main lines with 85-pound rails, using standard ties. Solid manganese-steel frogs are used throughout. Convenient passing tracks and spurs are located on all benches as close

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to the shovels as practicable in order to eliminate delays in spotting trains for loading. Wherever traffic is heavy on the main lines, double trackage is provided. The mileage of the present track system is as follows:

Main lines	11 miles
Bench tracks	12 miles
Dump tracks	5 miles
Total trackage	28 miles

The available motive power for the mine service consists of eleven 21 by 26 inch six-wheel American standard-gage locomotives weighing when coaled and watered approximately 180,000 pounds; eight 21 by 24 inch six-wheel Baldwin locomotives weighing approximately 170,000 pounds; and four smaller Porter locomotive weighing 95,000 pounds. The last are used for switching and the delivery of coal powder, etc. Coal and water for the locomotives are taken from the coaling station and water columns are located beyond the crushing plant at a point where all traffic must pass. Coal is loaded into the bunkers by gravity through chutes in the bottom of the bins. Water columns are so located that the locomotives can take water while being coaled.

Although several types of standard-gage automatic air-dump cars of 20 and 30 yard capacity are being used at present it is intended to standardize on the 20-yard drop-door dump cars for permanent equipment. The 30-yard cars, unless much more heavily constructed, are not rugged enough for the rough service required without undergoing excessive repairs; besides they are not best adapted either in length or capacity for use at the mine crushing plant.

When working five shovels per shift at least 14 locomotives are require for actual shovel-loading service, and not less than 110 cars. Two engines are required in switching and general pit service. Where grades are level or in favor of the loads, trains with 200 cubic yards of material are used, but where adverse grades exist 120 to 140 yard trains are in service.

The average haul for ore is 2.7 miles and for waste is 2.66 miles.

All trains are routed by dispatchers and switchtenders located at advantageous points in the pit and on the dumps, by means of signals and direct telephone connections. Semaphores are used wherever there is a possibility of congested traffic.

Fuel is delivered in railroad cars directly to the top of the locomotive coaling station. A locomotive crane with clamshell bucket transfers coal from the cars to the steam-shovel coal bunker.

WATER SUPPLY

The water supply is obtained by air-lift from wells drilled 800 to 2,000 feet deep. The water from three or four wells flows to a pumping station, which elevates it to steel storage tanks prior to distribution for domestic and industrial purposes. In addition some water from the underground development is pumped directly to a treatment plant. All industrial water is treated in

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three Reisert automatic lime-barium water-softening plants, each having a rated capacity of 6,000 gallons per hour. These plants are of such elevation that the entire mine is served by gravity. The raw water will average 18.04° permanent hardness and 9.3° temporary hardness.[3] The treated water averages 0.6° permanent hardness and 3.1° temporary hardness.

LIGHTING

The lighting for night work is done entirely with electricity. On the steam shovels 1-kilowatt generators driven by steam turbines are standard equipment for electric lighting. On the electric shovels current for lighting is taken directly from the power lines through transformers which step the voltage down to 125. Flood lights placed at convenient points on the shovel light both the pit and car which is being loaded.

PLANT

Complete machine, boiler, blacksmith, drill, car, carpenter, and electrical shops are provided for the maintenance of all equipment.

DRAINAGE

Drainage ditches have been constructed around the pits discharging into the flume and other channels, minimizing the flow of water into the pits. The quantity of water being pumped from the pits averages approximately 35 gallons per minute during most of the year. Unless the water interferes with the operation of a shovel the pits are only pumped out at such intervals as may be necessary to prevent any great accumulation. During the rainy season, which is about two and a half months of the year, the pits are pumped out every day. For normal pumping, bronze centrifugal pumps of a rated capacity of 500 gallons per minute are used. For emergency flood purposes three additional 1,200 gallons per minute bronze centrifugal pumps are available. These pumps are direct-connected to electric motors. Previous to the completion of the protective measures described, the emergency equipment was often in use for considerable periods, but under present conditions its use is seldom necessary. Two years before the completion of the flume all equipment operated 24 hours a day for three months.

SAFETY METHODS AND ORGANIZATION

A safety organization is maintained under a full-time safety inspector who makes daily inspection of equipment and is in constant touch with the foremen and men. Suggestions of employees and recommendations of the inspector are given prompt attention and made effective if found practical. All accidents, no matter

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3 - Temporary hardness is supposed to represent combined carbonates of lime and magnesia; permanent hardness is supposedly due to lime and magnesia in other forms. The French degree of hardness represents parts of calcium carbonate per 100,000; multiply by 10 to convert to parts per million. See Scott, W. W., Standard Methods of Chemical Analysis, p. 1440, 1922.

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how trivial, are thoroughly investigated and the causes determined, so that the necessary changes in equipment, methods, and placing of men can be made to prevent their recurrence. The Bureau of Mines first-aid and mine rescue car makes periodical visits.

The hospital and medical staff are centrally located and can respond within a few minutes to any emergency calls.

WAGE, CONTRACT, AND BONUS SYSTEMS

The Chino mines are operated on a straight wage and salary basis and no bonuses of any kind are paid.

ADMINISTRATIVE ORGANIZATION

The administrative and mine organization is shown by the following chart (fig. 14).

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Table 3. - Summary of costs of electric shovels, 1929

Cubic yards all material loaded	4,287,027
Dry tons of ore loaded	2,621,340
Cubic yards of waste loaded	2,976,357
Shovel-shifts operated	3,602

	All material, cost per cubic yard	Ore, cost per ton	Waste, cost per cubic yard
Shovel operation			
Operating labor:			
Engineers	$ 0.00713	$ 0.00386	$ 0.00687
Cranemen	.00423	.00265	.00375
Foremen	.00328	.00179	.00315
Pitmen	.00749	.00375	.00749
Other operating labor	.00370	.00237	.00324
Total operating labor	$ 0.02583	$ 0.01442	$ 0.02450
Operating supplies:			
Fuel or power	$ 0.01121	$ 0.00517	$ 0.01160
Grease and lubricants	.00057	.00030	.00057
Other operating supplies	.00072	.00031	.00076
Total operating supplies	$ 0.01250	$ 0.00578	$ 0.01293
Shovel repair:			
Shovel repair labor	$ 0.00255	$ 0.00143	$ 0.00241
Shop labor	.00519	.00216	.00557
Repair supplies	.01892	.00972	.01869
Shovel track and general overhauling	.01137	.00253	.01414
Total shovel repairs	$ 0.03803	$ 0.01584	$ 0.04081
Total shovel operation	$ 0.07636	$ 0.03604	$ 0.07824
Drilling			
Operating labor	$ 0.02645	$ 0.01334	$ 0.02635
Air	.00387	.00199	.00382
Operating supplies	.00274	.00145	.00268
Repair labor	.00341	.00172	.00340
Repair supplies	.00514	.00256	.00514
Total drilling	$ 0.04161	$ 0.02106	$ 0.04139
Blasting			
Labor	$ 0.00641	$ 0.00375	$ 0.00593
Powder	.04378	.02126	.04434
Other supplies	.00489	.00247	.00487
Total blasting	$ 0.05508	$ 0.02748	$ 0.05514
Haulage			
Locomotives	$ 0.14229	$ 0.07715	$ 0.13701
Cars	.02999	.01375	.03109
Track maintenance	.03046	.01704	.02886
Dump maintenance	.01548	---	.02229
Total haulage	$ 0.21822	$ 0.10794	$ 0.21925
General charges	$ 0.02943	$ 0.01432	$ 0.02977
Grand total	$ 0.42070	$ 0.20684	$ 0.42379

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Table 4. - Summary of costs of steam shovels, 1929

Cubic yards all material loaded	2,245,111
Dry tons of ore loaded	1,335,476
Cubic yards of waste loaded	1,577,373
Shovel-shifts operated	2,908

	All material, cost per cubic yard	Ore, cost per ton	Waste, cost per cubic yard
Shovel operation			
Operating labor:			
Engineers	$ 0.01067	$ 0.00484	$ 0.01108
Cranemen	.00839	.00377	.00876
Firemen	.01089	.00475	.01148
Foreman	.00530	.00241	.00549
Pitmen	.01567	.00675	.01659
Other operating labor	.01080	.00411	.01190
Total operating labor	$ 0.06172	$ 0.02663	$ 0.06530
Operating supplies:			
Fuel or power	$ 0.04464	$ 0.01951	$ 0.04701
Water	.00278	.00123	.00291
Grease and lubricants	.00097	.00045	.00100
Coal crane expense	.00901	.00399	.00945
Other operating supplies	.00095	.00041	.00101
Total operating supplies	$ 0.05835	$ 0.02559	$ 0.06138
Shovel repair:			
Shovel repair labor	$ 0.00772	$ 0.00309	$ 0.00837
Shop labor	.00595	.00238	.00646
Repair supplies	.02428	.00982	.02625
Shovel track and general overhauling	.02436	.01165	.02481
Total shovel repairs	$ 0.06231	$ 0.02694	$ 0.06589
Total shovel operation	$ 0.18238	$ 0.07916	$ 0.19257
Drilling			
Operating labor	$ 0.02686	$ 0.01342	$ 0.02687
Air	.00404	.00201	.00404
Operating supplies	.00273	.00127	.00281
Repair labor	.00343	.00170	.00344
Repair supplies	.00507	.00255	.00506
Total drilling	$ 0.04213	$ 0.02095	$ 0.04222
Blasting			
Labor	$ 0.01025	$ 0.00450	$ 0.01078
Powder	.04722	.02264	.04805
Other supplies	.00598	.00270	.00622
Total blasting	$ 0.06345	$ 0.02984	$ 0.06505
Haulage			
Locomotives	$ 0.18142	$ 0.09623	$ 0.17675
Cars	.03092	.01417	.03202
Track maintenance	.04629	.02011	.04885
Dump maintenance	.01848	---	.02630
Total haulage	$ 0.27711	$ 0.13051	$ 0.28392
General charges	$ 0.04450	$ 0.01724	$ 0.04874
Grand total	$ 0.60957	$ 0.27770	$ 0.63250

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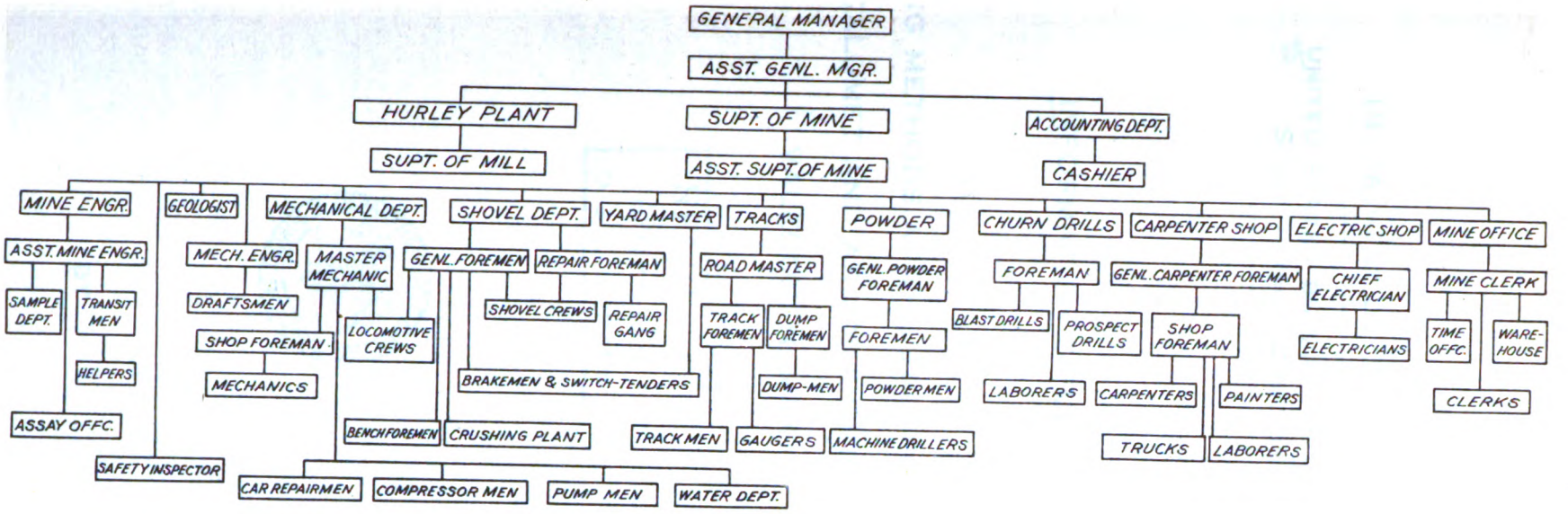

Figure 14.-Organization chart

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